I. C. 6479

July, 1931

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
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MILLING METHODS AND COSTS AT THE ARTHUR AND MAGNA CONCENTRATORS OF THE UTAH COPPER CO.

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MILLING METHODS AND COSTS

AT THE ARTHUR AND MAGNA CONCENTRATORS OF THE UTAH COPPER CO.[1]

By H. S. Martin[2]

INTRODUCTION

This paper, describing the concentrating practices at the plants of the Utah Copper Co., is one of a series being prepared by the United States Bureau of Mines in which the data are given in as nearly uniform a manner as possible.

The Magna mill is 19 miles west of Salt Lake City on the routes of the Lincoln and Victory highways, and the Arthur plant is 1 mile west of the Magna mill. Both are on the main road west of the town of Magna and east of the town of Garfield.

ACKNOWLEDGMENTS

Thanks are due to D. C. Jackling, president, and D. D. Moffat, vice-president and general manger, of the Utah Copper Co., for their permission to prepare and publish this paper.

Some of the data used in this paper on microscopic examinations of Utah Copper mill products, and the tables showing screen sizes and total recoveries of copper after coarse and fine grinding operations have been published in Technical Paper 255 of the American Institute of Mining and Metallurgical Engineers, by the same author. The microscopic work was done by R. E. Head and A. L. Crawford of the cooperative microscopic laboratory of the United States Bureau of Mines at the University of Utah.

Figure 4, showing the old and new ball-mill grinding circuits at both Arthur and Magna mills, was presented in a paper by J. V. N. Dorr and A. D. Marriott of the Dorr Co., at the San Francisco meeting of the American Institute of Mining and Metallurgical Engineers, October 8, 1929.

HISTORY OF MINE AND CONCENTRATING PLANTS

The mine of the Utah Copper Co. is in Bingham Canyon in the Oquirrh Range of mountains, 17 miles south of the mill sites. The deposit of low-grade ore was discovered in the 70's, but the claims were abandoned until relocated in 1888 by Col. E. A. Wall. Colonel Wall interested Capt. J. R. DeLamar in the property, and through DeLamar D. C. Jackling and R. C. Gemmell examined the property in 1898.

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[1] - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6479."

[2] - Metallurgical engineer, Arthur concentrator, Utah Copper Co., and one of the consulting engineers, U. S. Bureau of Mines.

In 1903 Jackling, after many other unsuccessful negotiations, obtained the financial backing of C. M. McNeil, Spencer Penrose, and R. A. F. Penrose, and the Utah Copper Co. was organized. The original estimate of the ore contained in the mine amounted to 25,000,000 tons averaging 2 per cent of copper. In the summer of 1903 construction work was started on a pilot mill at Copperton in Bingham Canyon about 2-1/2 miles below the mine. Methods of concentrating the ore were worked out in this pilot plant, and it operated from 1905 to 1910. Operations were economically a success from the start; the company gradually expanded, acquiring more ground at the mine, continuing churn-drill prospecting and increasing the capacity of the mills until at the end of 1929 ore reserves were estimated at 640,000,000 tons averaging 1.07 per cent of copper, even though from 1903 to 1929 more than 200,000,000 tons of ore had been mined and concentrated.

The Magna plant, the original large-scale Utah Copper mill, was first put into operation in 1907, and gravity methods of concentrating were used. The Arthur plant, originally built by the Boston Consolidated Mining Co., was purchased by the Utah Copper Co. in 1910.

When the Magna mill was first completed, its daily capacity was rated at 6,000 tons, and in 1909 the daily capacity of the Arthur plant was rated at 3,000 tons. Both mills have been enlarged and changed, until each can now handle more than 30,000 tons of ore per day, although for best metallurgical results a total daily tonnage of 54,000 is probably the upper limit. A record of 69,200 dry tons of ore handled in one 24-hour period was made during 1929.

Before 1914, gravity concentrating methods were practiced exclusively, and the machines used comprised jigs, vanners, and tables. From 1914 to 1926 a combination of gravity and flotation methods was used, and since 1926 concentration has been by flotation methods exclusively.

Water supply and tailings disposal were the two factors that decided the location of the large-scale concentrators. Springs near the Magna plant were first used for water supply. The Utah Copper Co. later bought an interest in the Utah & Salt Lake Canal Co., which owned an irrigation canal running from Utah Lake out into the farming country between Salt Lake City and Magna. This canal was enlarged and extended to the Magna plant, and water is delivered by gravity from Utah Lake to an elevation only 26 feet lower than that of the Magna mill reservoir. From here it is pumped to the mill reservoirs at Magna and Arthur.

Water used for culinary purposes at the Garfield town site and for the houses at the mills is obtained from artesian wells 3 miles east of the Magna mill site.

Originally a steam-electric power plant was built at the Magna plant to furnish power for the mill, but this was later abandoned and for a good many years all power has been obtained from the hydroelectric plants of the Utah Power & Light Co. At the present, with normal operating conditions, the Utah Copper Company's load from this source averages about 50,000 hp., approximately one-third of the total power output of the power company.

MINING METHODS AND ORE TREATED

Mining is done entirely by electric shovels from an open-cut mine, the mountain being terraced into levels with differences in elevation of 50 to 60 feet. Trains of 6, 8,

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or 10 standard-gage railroad cars of 80 to 100 tons capacity are loaded on the various levels by the shovels and hauled to the yard by steam and electric locomotives, where trains of 50 cars are made up for transportation to the mills. The 50-car trains are transported 17 miles to the mills by Mallet steam locomotives over the main line of the Bingham & Garfield Railroad.

Although the bulk of the ore treated at the concentrators is monzonite porphyry containing finely disseminated iron and copper sulphides, many other kinds of rock occur in the mine in fairly large quantities. The quartzite is sufficiently mineralized for a distance of 300 feet from the porphyry contact to be called ore; an extensive body of epidotized limestone is also being mined as ore. The gangue rock in the upper levels of the mine is higher in silica and lower in alumina contents than the porphyry of the lower levels. The following chemical analyses give an idea of the variation in types of rock:

Analyses, per cent	Types of rock			
	Dark porphyry	Green ore J level	Quartzite	Light porphyry
Fe_2O_3	4.05	7.70	Trace	0.30
Al_2O_3	15.15	9.60	4.45	14.30
CaO	Trace	11.20	Trace	Trace
MgO	2.15	14.60	.45	1.15
S	1.45	.75	7.85	1.35
Cu	.59	.20	1.03	2.01
SiO_2	62.70	53.90	79.30	68.80
TiO_2	2.50	None	None	Trace
P_2O_5	.40	.35	Trace	.15
$K_2O + Na_2O$	8.55	.95	Trace	10.90
Fe as sulphide	1.10	.50	6.85	.65

The sulphide minerals present in the ore are pyrite, chalcopyrite, chalcocite, covellite, and bornite. A monthly composite sample of mill heads when examined under the microscope showed the following percentages of the various sulphide minerals:

	Per cent by weight of the original ore	Per cent by weight of total sulphide copper minerals
Pyrite	2.60	-
Chalcopyrite	1.71	78.0
Chalcocite	.21	9.6
Covellite	.17	7.6
Bornite	.11	4.8
Total sulphide copper	.932	-
Total nonsulphide copper	.052	-

The samples used for the mineral analysis given in the preceding table were obtained from a composite sample that represented nearly all parts of the mine. Mineral analyses of samples taken in different localities of the mine show marked variations, however, as compared to the analysis given.

Extensive data on the geology and the mineralogy of the Utah Copper ore deposit are given in The Disseminated Copper Ores of Bingham, Utah, published in the Transactions of the American Institute of Mining Engineers, volume 54, 1916.

Other geological data are given in The Ore Deposits of Utah, Professional Paper 111 of the U. S. Geological Survey, published in 1920. Quoting from page 360 of this paper:

> Underlying the zone of oxidation is the zone of secondary sulphides. In this zone the primary sulphides, pyrite and chalcopyrite and possibly some bornite, have been partly or wholly replaced by the copper sulphides covellite and chalcocite..........Beeson states that covellite is the more abundant secondary sulphide in the higher portions of the orebody but that chalcocite becomes increasingly abundant near the bottom of the canyon..........Underlying the zone of secondary sulphides is that of primary sulphides. Much of the material that contains little covellite or chalcocite is ore, but it decreases in tenor with increase in depth. This naturally suggests enrichment by the addition of copper in the form of chalcopyrite, and this seems to be the theory held by those most familiar with the ore deposit. Beeson has shown that some of the material that appears to be chalcopyrite is only a thin film of that material covering covellite or chalcocite, all three of the sulphides being of secondary origin.

Figure 1 shows six photomicrographs of mineral associations occurring in Utah Copper mill products. These photomicrographs are lettered A to F inclusive, and a description of each follows.

A - Pyrite crystals incased in thin chalcocite shells

The pyrite is dark because of the rough, unpolished condition of its surface. The chalcocite surface was polished smooth so that it reflected a large amount of light to the photographic plate and therefore appears light in the picture. Thin shells of chalcocite and covellite often occur surrounding pyrite grains in the plus 100 and plus 150 mesh sizes but are difficult to show in a photograph owing to the great difference in the hardness of the minerals and to the fact that chalcocite and covellite, unless highly polished, appear similar to the bakelite matrix in a photograph. The thickness of the chalcocite shell is 10 to 25 microns. It is evident that pyrite grains coated as illustrated react with flotation reagents as though they were entirely chalcocite, and until the shell is removed by finer grinding a clean separation is impossible. These grains are from the plus 150-mesh size Arthur concentrates.

B - Pyrite imbedded in chalcopyrite and covellite

From a briquet of Arthur mill concentrates minus 100-mesh plus 150-mesh sizes. Black spots represent pits which did not reflect light to the photographic plate. The theory

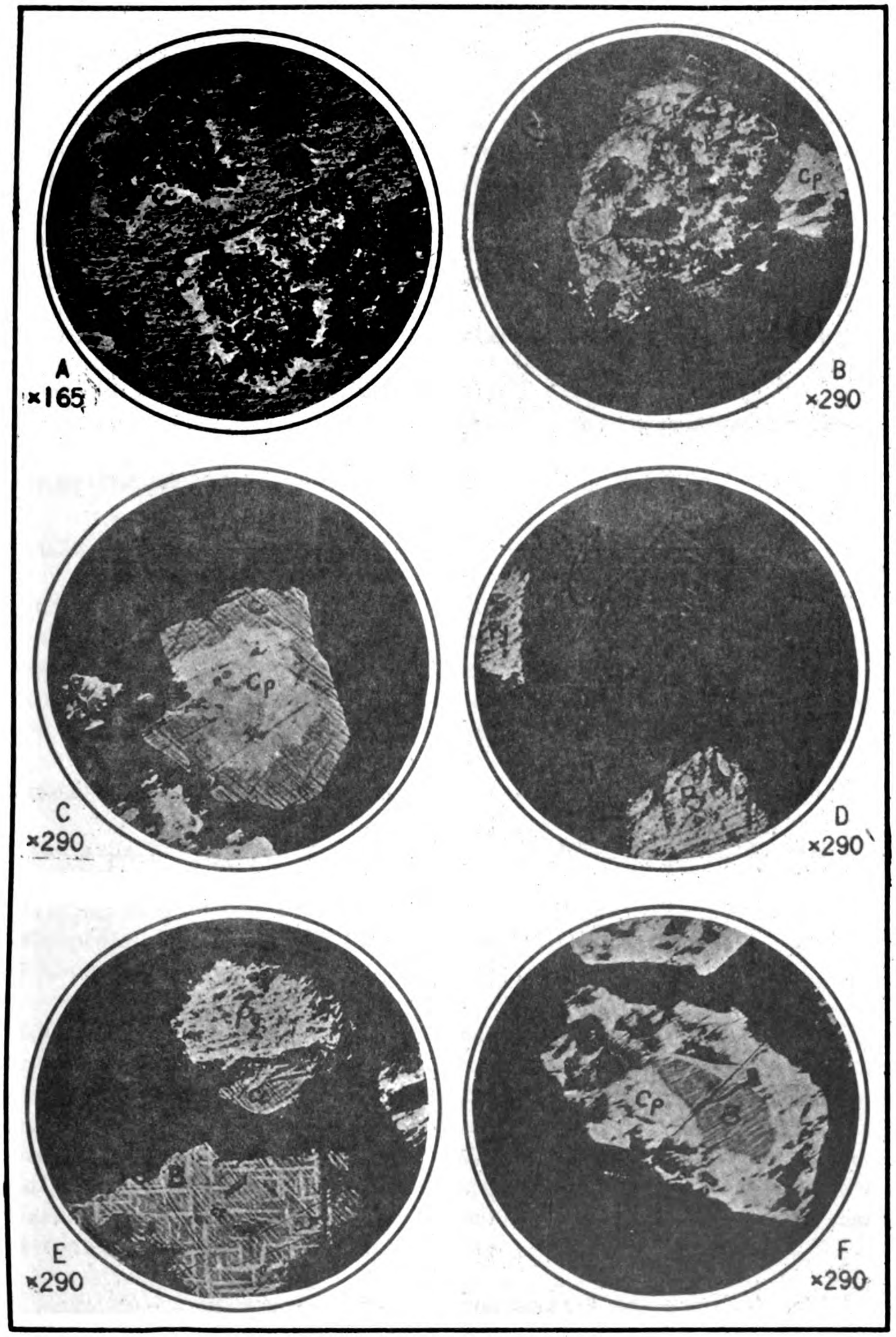


Figure 1.- Photomicrographs of mineral associations: Cc, Chalcocite; Cp, chalcopyrite; Py, pyrite; B, bornite; and Cv, covellite

advanced in the description of A regarding floatability and liberation applies to grains of this character.

C - Chalcopyrite incased in chalcocite

From Arthur concentrates minus 100-mesh plus 150-mesh sizes. Although this grain is composed of two minerals liberation is not as important as with the pyrite-copper minerals shown in the preceding figure. Presumably, this grain would have the floatability of chalcocite, and finer grinding, liberating the constituent minerals, would be of advantage because under prevailing practice at the Utah Copper Co. plants higher recovery is made of chalcopyrite than of chalcocite.

D - A free pyrite grain, a pyrite cube surrounded by chalcocite, and a grain of bornite chalcocite

An area on the surface of a briquet of Arthur concentrates minus 100-mesh plus 150-mesh sizes. Shells of copper sulphides on pyrite grains are undoubtedly responsible for presence of some of the pyrite in the concentrates.

E - Complex bornite-chalcopyrite association

Arthur concentrates, minus 100-mesh plus 150-mesh sizes, showing bornite containing secondary chalcopyrite veinlets arranged along the crystallographic directions of the bornite. This figure illustrates the impossibility of securing complete liberation of copper minerals even by extremely fine grinding. In the pyrite-chalcocite middlings grain the intergrowth is not so intimate and would be broken up by slightly finer grinding.

F - Bornite-chalcopyrite-covellite intergrowth

Particle from Arthur mill concentrates, minus 100-mesh plus 150-mesh sizes. The center is a core of bornite about 35 microns thick containing minute secondary veinlets of chalcopyrite, surrounded by a rim of chalcopyrite about 30 microns thick, which in turn is incased in a thin shell of covellite 7 to 10 microns thick. Black spots represent pits in the surfaces of the minerals.

Certain areas of the mine had been opened up by tunnels and crosscuts a good many years ago, and some high-grade ore had been stoped out and shipped before milling operations commenced. The areas in which these stopes occur have always yielded ore known as "bad ore" to flotation operators, due to oxidation of the minerals and weathering of the gangue constituents. Operations in the main ore body have now been carried so far back into the mountain that most of the ore surrounding the old stopes has been removed. This has resulted in a gradual decrease in the nonsulphide copper content of mill heads, which, of course, means that the ore is constantly becoming more amenable to treatment by flotation.

HISTORY OF CONCENTRATOR OPERATIONS

The original coarse crushing machines at both the Arthur and Magna plants were No. 6 and No. 7-1/2 gyratory crushers, the rock being dumped onto grizzlies from bottom-dump cars and the larger rocks being broken by hand to pass the grizzlies. In 1919 an electric car dumper and a No. 27 gyratory crusher were installed at Arthur, and in 1923 a twin car dumper and a No. 27 gyratory were put in at Magna. The second stage of crushing at the Arthur plant was by Nissen stamps and at the Magna plant by 54 by 20 inch Garfield rolls. After the Utah Copper Co. took over the Boston Consolidated mill the rolls were put in at the Arthur plant and the stamps abandoned.

The only equipment originally used in the Magna mill and still in use since 1910 at the Arthur mill is the 37-1/2 by 15 inch Garfield fine rolls, which reduce the ore from 1-inch to 8-mesh size. Some of these rolls have been replaced by the larger 44 by 16 inch size. Chilian mills were formerly used at both mills following the fine rolls, but were eliminated entirely in 1924, ball mills having proved more efficient.

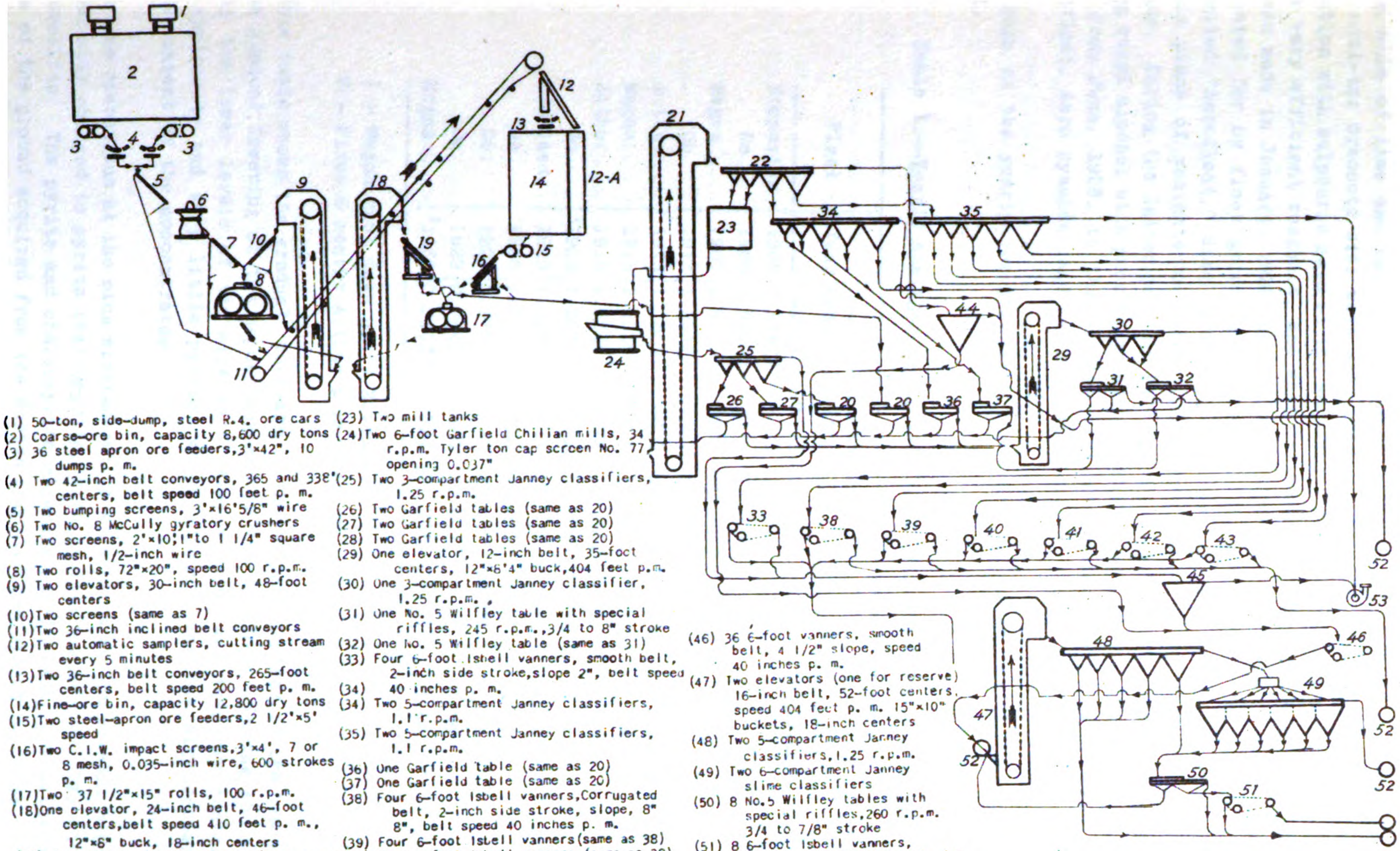
Up to 1929 the No. 27 gyratory crusher product at both plants passed through No. 7-1/2, No. 8, or No. 9 gyratory crushers, where it was reduced from 6-inch to 3-inch maximum size, then through two stages of 74 by 20 or 54 by 20 inch Garfield rolls in closed circuit with screening towers, the screen undersize product containing approximately the same sizes as the Symons crusher product shown in Table 3. In 1929 four 7-foot Symons cone crushers were installed at each plant, and these crushers reduce the 6-inch material in one pass to from 3/4 to 1 inch size, thus eliminating 9 small gyratory crushers and 14 sets of large rolls.

The feed to the concentrating department was first classified in Traylor iron classifiers and the products distributed to Traylor jigs, Wilfley tables, Johnson slime vanners, and Johnson sand vanners. The Traylor jigs were soon replaced by Garfield roughing tables and the Traylor classifiers by Richards-Janney hydraulic classifiers. With the advent of ball-mill grinding the hydraulic classifiers were gradually replaced by Dorr drag classifiers, and finally Dorr bowl-drag classifiers were added during the last year.

Figure 2 gives a gravity-concentration flow sheet of the Arthur plant in 1911 and 1912 and is typical of both plants at the time. This figure, compared with Figures 3 and 4 which give present flow sheets, shows the almost complete changes in equipment made since that date.

Flotation was first used at the Utah copper plants in 1914 to grade up the siliceous low-grade concentrates. Work also was done from this date on in the treatment of slimes feed by flotation. The finer portions of the ore, formerly slimes vanner feed, had always been the cause of poor mill recoveries, and the successful treatment of this material offered a large margin of additional recovery. A fairly successful flotation treatment was worked out by 1916, and work commenced on the erection of a large "slimes" flotation plant at the Arthur mill. This plant was in full operation by May, 1918, treating what was formerly the slimes vanner-feed product. The slimes vanners were then discarded. The sand vanners were also eliminated in 1920, and the entire reground tailings products from the Garfield and Wilfley tables were treated by flotation methods. The Magna plant was shut down early in 1919 and was remodeled, before starting up in November, 1922, to handle feed on Garfield roughing and Wilfley finishing tables, followed by flotation for the combined reground table tailings products. At both plants the Garfield and Wilfley tables were eliminated in 1926, and the crude ore was treated entirely by flotation methods.

The reagents used in the treatment of vanner low-grade concentrates were "reconstructed" coal-tar creosote oil, heavy petroleum oils, and calura. The coal-tar creosote oils were "reconstructed" by heating with sulphur, and calura, an alkaline sulphide, was made



(1) 50-ton, side-dump, steel R.4. ore cars
(2) Coarse-ore bin, capacity 8,600 dry tons
(3) 36 steel apron ore feeders, 3'×42", 10 dumps p. m.
(4) Two 42-inch belt conveyors, 365 and 338' centers, belt speed 100 feet p. m.
(5) Two bumping screens, 3'×16' 5/8" wire
(6) Two No. 8 McCully gyratory crushers
(7) Two screens, 2'×10'; 1" to 1 1/4" square mesh, 1/2-inch wire
(8) Two rolls, 72"×20", speed 100 r.p.m.
(9) Two elevators, 30-inch belt, 48-foot centers
(10) Two screens (same as 7)
(11) Two 36-inch inclined belt conveyors
(12) Two automatic samplers, cutting stream every 5 minutes
(13) Two 36-inch belt conveyors, 265-foot centers, belt speed 200 feet p. m.
(14) Fine-ore bin, capacity 12,800 dry tons
(15) Two steel-apron ore feeders, 2 1/2'×5' speed
(16) Two C.I.W. impact screens, 3'×4', 7 or 8 mesh, 0.035-inch wire, 600 strokes p. m.
(17) Two 37 1/2"×15" rolls, 100 r.p.m.
(18) One elevator, 24-inch belt, 46-foot centers, belt speed 410 feet p. m., 12"×8" buck, 18-inch centers
(19) Four C.I.W. impact screens (same as 16
(20) Six Garfield tables, 4'×12', 245 r.p.m 7/8-inch stroke
(21) One elevator, 30-inch belt, 58-foot centers, 15"×10" buck
(22) Four 4-compartment Janney classifiers, 1.25 r.p.m.
(23) Two mill tanks
(24) Two 6-foot Garfield Chilian mills, 34 r.p.m. Tyler ton cap screen No. 77 opening 0.037"
(25) Two 3-compartment Janney classifiers, 1.25 r.p.m.
(26) Two Garfield tables (same as 20)
(27) Two Garfield tables (same as 20)
(28) Two Garfield tables (same as 20)
(29) One elevator, 12-inch belt, 35-foot centers, 12"×6'4" buck, 404 feet p.m.
(30) One 3-compartment Janney classifier, 1.25 r.p.m. ,
(31) One No. 5 Wilfley table with special riffles, 245 r.p.m., 3/4 to 8" stroke
(32) One No. 5 Wilfley table (same as 31)
(33) Four 6-foot Isbell vanners, smooth belt, 2-inch side stroke, slope 2", belt speed
(34) 40 inches p. m.
(34) Two 5-compartment Janney classifiers, 1.1 r.p.m.
(35) Two 5-compartment Janney classifiers, 1.1 r.p.m.
(36) One Garfield table (same as 20)
(37) One Garfield table (same as 20)
(38) Four 6-foot Isbell vanners, Corrugated belt, 2-inch side stroke, slope, 8" 8", belt speed 40 inches p. m.
(39) Four 6-foot Isbell vanners (same as 38)
(40) Four 6-foot Isbell vanners (same as 38)
(41) Four 6-foot Isbell vanners (same as 38)
(42) Four 6-foot Isbell vanners (same as 38)
(43) Four 6-foot Isbell vanners (same as 38)
(44) 12 cone settlers
(45) 16 cone settlers
(46) 36 6-foot vanners, smooth belt, 4 1/2" slope, speed 40 inches p. m.
(47) Two elevators (one for reserve) 16-inch belt, 52-foot centers, speed 404 feet p. m. 15"×10" buckets, 18-inch centers
(48) Two 5-compartment Janney classifiers, 1.25 r.p.m.
(49) Two 6-compartment Janney slime classifiers
(50) 8 No. 5 Wilfley tables with special riffles, 260 r.p.m. 3/4 to 7/8" stroke
(51) 8 6-foot Isbell vanners, corrugated belts, 2-inch side stroke 4-inch slope, speed 64 inches p. m.
(52) Automatic samplers, cutting stream every 5 minutes
(53) Four 14-inch centrifugal pumps, 5,000 gals. p. m.

Figure 2.- Flow sheet of the Arthur concentrator as operated in 1912

by boiling milk of lime and caustic soda with sulphur. From 1916 to 1926 these "reconstructed" coal-tar creosote oils were also used in flotation treatment of the "slimes" feed in conjunction with sulphuric acid as the modifier. These reconstructed oils, with sulphuric acid, were very efficient reagents, and when the change to the xanthate, pine oil, and lime reagents was made in January, 1926, no change in metallurgical results was obtained, except that accounted for by finer grinding and better classification. In January, 1927, a new reagent called "Aerofloat," discovered and developed by the research department, was put into use in place of xanthate and pine oil, this reagent being used in conjunction with Aero cyanide. During the latter part of 1927 and up to June, 1928, the new reagent was made by treating ethyl alcohol with phosphorus pentasulphide, and raw cresylic acid was used as the frother. From June, 1928, to the present time the reagents used have been lime, sodium ethyl Aerofloat, Aero cyanide, and raw cresylic acid.

Some of the yearly results since the start of operations at Copperton are given in Table 1.

Table 1.—Yearly analyses of concentrator feed and final products

Plant	Year	Analyses, per cent				
		Heads, copper	Tailings copper	Concentrates		
				Copper	Iron	Insoluble
Copperton..	1905	1.9787	0.7095	31.7786	-	-
Do.	1906	1.9607	.8223	27.2180	-	-
Magna..........	1907	1.8176	.6508	28.8890	14.532	30.083
Do.	1913	1.3013	.4942	18.2770	18.208	36.371
Arthur........	1913	1.1809	.4656	16.1660	21.108	32.058
Magna..........	1918	1.2026	.5154	15.6833	28.331	18.145
Arthur........	1918	1.2545	.3622	16.4620	24.489	23.805
Do.	[1]1922	1.2575	.2392	23.478	22.702	18.810
Combined....	1925	1.0225	.1443	17.475	28.44	16.14
Do.	1926	1.0102	.1325	17.071	28.63	15.05
Do.	1927	0.9787	.1222	25.197	26.58	10.27
Do.	1929	0.9945	.1480	32.064	25.81	6.36
Magna..........	[2]1930	0.9603	.1114	31.762	26.28	5.44

1 - Magna shut down until November.
2 - First 6 months Arthur plant was not operated.

This table shows the gradual decrease in the copper content of the heads handled and also the gradual lowering of the copper content in the tailings. During the years 1905 to 1910 only the lower levels of the mine were being worked, and this ore contained predominantly chalcocite and very little pyrite, which accounts for the high copper content and low iron content of the concentrates.

As the operations at the mine widened out to a larger area the average mineral composition gradually changed to pyrite chalcopyrite, with smaller amounts of chalcocite, bornite, and covellite. The pyrite and chalcopyrite minerals particularly predominated in the upper levels of the ground acquired from the Boston Consolidated Mining Co. For this reason the copper content of the concentrates decreased gradually to 1918, then increased again with the advent of slimes flotation, due to lowering of the insoluble content. From 1922 to 1926 the grade of concentrates decreased again, due to increasing pyrite and chalcopyrite, then rose sharply in 1927, when the depression of pyrite was first achieved in flotation. The new reagents, used since 1927, have had the effect of eliminating both pyrite and insoluble matter from the concentrates.

The reduction in tailings losses since 1918 has been due primarily to better classification and finer grinding and second to the change for the better in the character of ore mined, as noted before. The present-day reagents are faster than the "reconstructed" oil and acid reagents and permit larger tonnages to be handled in a given machine, thus lowering power requirements in the flotation operation, but the final tailings made by the older reagents were probably just as low as can be made on the same ore at present. The increase in tailings losses for 1929 was caused by extremely high tonnages carried by the mills during the first five months of the year, before new grinding equipment had been installed. For the last seven months of the year, with lower tonnages and with 31 new ball mills in operation, the average tailings contained 0.11 per cent of copper.

The effect of flotation treatment of tailings losses is shown first in the comparative yearly results for Arthur and for Magna in 1918, when the Arthur "slimes" flotation plant was in operation and the Magna plant was treating the slimes product on vanners.

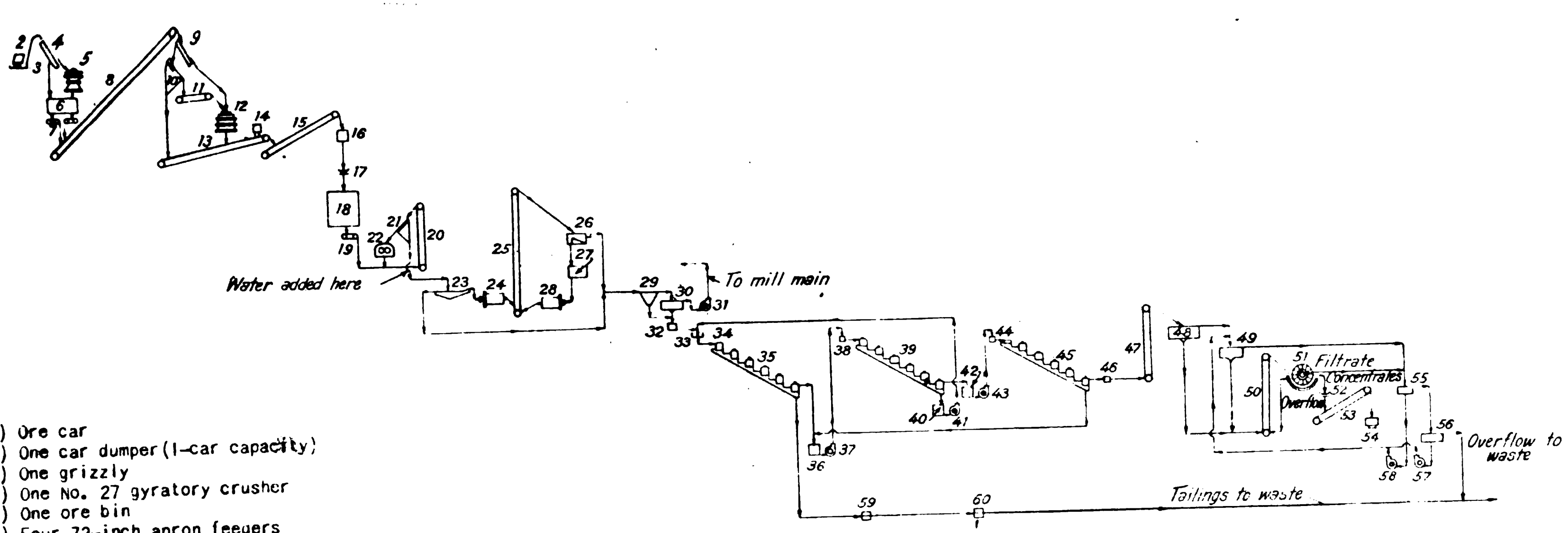
Table 2, which is taken from the annual report of the Utah Copper Co. for 1929, gives comparative cost data on all operations for the past 20 years. Quoting from the report:

> "The per-ton operating costs on concentrating ore, including all fixed, general and maintenance charges (except Federal taxes) are shown by periods from 1910 to 1929 inclusive. The increased total costs for 1929 as compared to the previous year were occasioned almost wholly by higher wage scales prevailing in the latter year."

Table 2

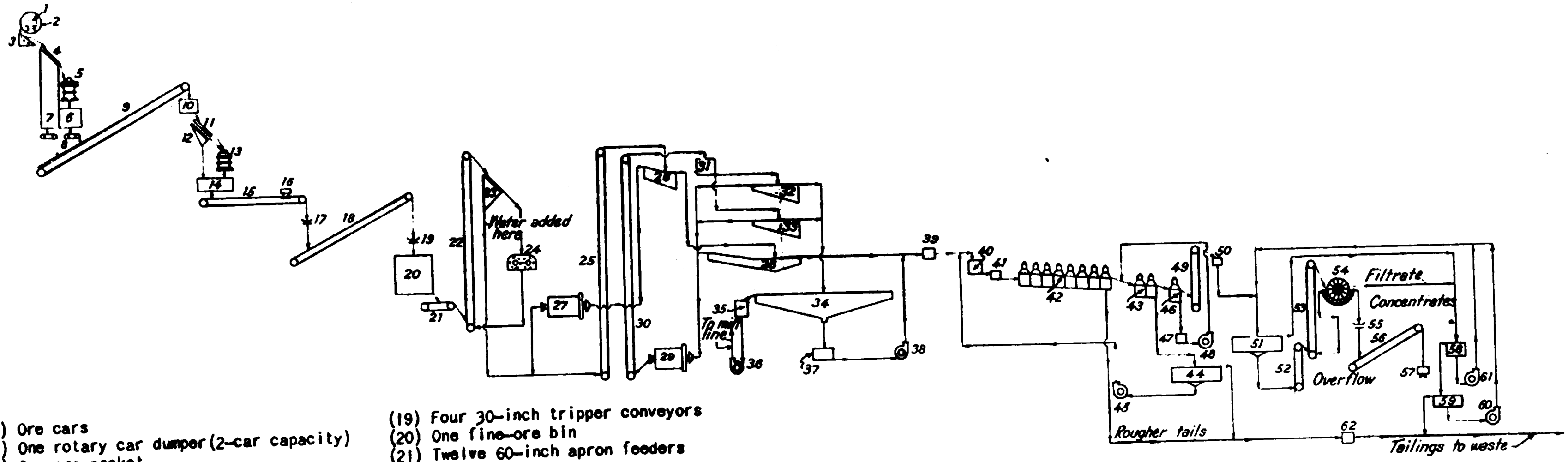
Years	Tonnage	Mining	Ore delivery	Milling	Total
1910-1916	47,814,225	$0.3308	$0.2841	$0.3824	$0.9973
1917-1923	52,550,951	.4506	.2361	.8661	1.5528
1924-1926	52,356,500	.3460	.1093	.5153	.9706
1927	13,811,500	.3734	.0926	.4523	.9183
1928	16,558,500	.3178	.0874	.3690	.7742
1929	17,724,100	.4121	.0896	.3658	.8675

During the first few years of flotation treatment the cost of this operation was of course much higher than the cost of gravity concentration, partly due to the cost of reagents and partly to the finer grinding and power involved. A gradual decrease in the cost of milling has been achieved since 1924, however, by simplification and increase in tonnage, in spite of the fact that the cost of grinding to the fineness now considered necessary has increased enormously over the cost of this item some years ago.



(2) Ore car
(3) One car dumper (1-car capacity)
(4) One grizzly
(5) One No. 27 gyratory crusher
(6) One ore bin
(7) Four 72-inch apron feeders
(8) Two 60-inch conveyors
(9) Two grizzlies
(10) Four impact screens
(11) Four 42-inch conveyors
(12) Four 7-foot Symons cone crushers
(13) Two 36-inch conveyors
(14) Two weightometers
(15) Two 36-inch conveyors
(16) Two head samplers
(17) Two 32-inch tripper conveyors
(18) One fine-ore bin
(19) 26 60-inch apron feeders
(20) Thirteen 30-inch elevators
(21) 78 impact screens
(22) Fourteen 42×16" rolls, twelve 37 1/2×15" rolls
(23) 26 15-foot Dorr bowl classifiers
(24) Thirteen 7×10' ball mills
(25) Thirteen 36-inch elevators
(26) 36 12×16' drag classifiers
(27) 39 sand boxes
(28) 39 7×10' ball mills
(29) Four sand boxes
(30) Thirteen 75-foot Dorr tanks
(31) Four 14-inch pumps
(32) Four 15×16' tanks
(33) Two feed distributing launders
(34) 99 rougher emulsifier cells
(35) 515 rougher flotation cells
(36) Two sumps
(37) Six 6-inch pump
(38) One distributing box
(39) 66 primary cleaner flotation cells
(40) One sump
(41) Three 6-inch pumps
(42) One sump
(43) Three 6-inch pumps
(44) One distributing box
(45) 35 secondary cleaner flotation cells
(46) One concentrates sampler
(47) Three 36-inch elevators
(48) Two 100-foot Dorr tanks
(49) Four 75-foot Dorr tanks
(50) Two 18-inch elevators
(51) Five American filters
(52) One 42-inch conveyor
(53) One 30-inch conveyor
(54) Concentrates cars to smelter
(55) Old concentrates bin used for settling
(56) One settling pond
(57) One 6-inch pump, intermittent service
(58) One 6-inch pump (same as 57)
(59) One tailings sampler
(60) One general tails sampler

Figure 3.- Flow sheet of the Arthur mill



(1) Ore cars
(2) One rotary car dumper (2-car capacity)
(3) One ore pocket
(4) Two grizzlies
(5) One No. 27 gyratory crusher
(6) Two hoppers
(7) One storage bin
(8) Four 60-inch apron feeders
(9) Two 54-inch conveyors
(10) One hopper
(11) Four grizzlies
(12) Four impact screens
(13) Four 7-foot Symons cone crushers
(14) One bin
(15) Two 54-inch conveyors
(16) Two weightometers
(17) Two 36-inch conveyors
(18) Two 42-inch conveyors
(19) Four 30-inch tripper conveyors
(20) One fine-ore bin
(21) Twelve 60-inch apron feeders
(22) Twelve 36-inch elevators
(23) 108 impact screens
(24) 36 42×16" rolls
(25) Twelve 36-inch elevators
(26) Eighteen 8×14' drag classifiers
(27) Eighteen 7×10' ball mills
(28) Twelve 22-foot Dorr bowl classifiers
(29) 36 7×10' ball mills
(30) Twelve 36-inch elevators
(31) 12 mechanical feed distributors
(32) 36 6'×17'6" drag classifiers
(33) 36 8'×17'6" drag classifiers
(34) One 225-foot Dorr thickener
(35) One sump
(36) Three 14-inch pumps
(37) One sump
(38) Five 8-inch pumps (Wilfley)
(39) Two samplers
(40) Two 5-way distributors
(41) Ten 6-way distributors
(42) 480 rougher flotation cells
(43) 120 primary cleaner flotation cells
(44) Six 75-foot Dorr tanks
(45) Four 6-inch pumps
(46) 60 secondary cleaner flotation cells
(47) Two sumps
(48) Four 6-inch pumps
(49) Four 36-inch elevators
(50) Two concentrates samplers
(51) Two 75-foot Dorr tanks
(52) One 18-inch booster elevator
(53) Two 18-inch elevators
(54) Five American filters
(55) One 30-inch conveyor
(56) One 30-inch conveyor
(57) Concentrates car to smelter
(58) Old concentrates bins
(59) Two settling ponds
(60) One 3-inch pump (intermittent service)
(61) One 4-inch pump (intermittent service)
(62) Two samplers

Figure 4.- Flow sheet of the Magna mill

PRESENT METHOD OF CONCENTRATING

Figures 3 and 4 give the most recent flow sheets of the Arthur and Magna mills, respectively. Each item of equipment in these flow sheets is numbered and sufficiently described in the accompanying legends to be self-explanatory. Further details of methods and equipment will be discussed by successive departments in the mills.

Dumping, primary and secondary crushing

The flow sheets of coarse crushing and secondary crushing at the two plants are practically the same, except that the Magna plant has a rotary car dumper handling two cars at a time, while the older dumper at the Arthur mill handles only one car and lifts the car to turn it over. Up to 1930 an electric mule was used at the Arthur plant to pull the ore cars up an incline onto the dumper platform, but the yard has now been rearranged, and a large earth fill made so that the electric yard locomotive can spot the cars on the dumper, the empty cars leaving the dumper by gravity, as has always been done at Magna. Each plant has one No. 27 Allis-Chalmers gyratory crusher, capable of taking rock 54 inches in diameter. Ahead of the crushers are grizzlies, spaced 5-1/2 inches, with bins under the grizzlies and hoppers under the crusher discharges. Steel apron feeders deliver both the undersize and crusher products to 54-inch inclined belts at Magna and 60-inch inclined belts at Arthur. The primary inclined belts discharge the ore over 2-1/2-inch grizzlies, the grizzly oversize going directly to the Symons crushers and the grizzly undersize passing over Duckworth impact screens with 1-inch openings. The impact-screen undersize goes to the fine-ore bins, and the oversize joins the 2 1/2-inch grizzly oversize going to the Symons crushers. The Symons product joins the impact-screen undersize going to the fine-ore bins. The primary and secondary crushing units at each plant can easily handle 35,000 tons of average ore per 24 hours with all four Symons crushers in operation, giving a total capacity of 70,000 tons per day.

With average ore, about 80 per cent of the material passes through the 5 1/2-inch grizzlies, and 20 per cent goes to the No. 27 crusher. Of the total tonnage approximately 17 per cent passes through the 1-inch impact screens and by-passes the Symons crushers.

The No. 27 crusher at the Arthur mill is driven by a 250-hp. 440-volt motor operated at 600 r.p.m. The motor is connected to the crusher pinion shaft through a single reduction-gear unit. The newer installation at Magna uses a 250-hp 440-volt motor operated at 300 r.p.m., direct-connected through a Duckworth flexible coupling to the crusher pinion shaft. The speed reduction ratio of each crusher from the pinion shaft to the eccentric is 2.88 to 1.

The Arthur car dumper uses two 162-1/2-hp., 250-volt D. C. motors operated at 350-r.p.m., geared through three reductions of spur gears to the cradle and equipped with magnetic brakes. The Magna plant rotary car dumper is driven by a 50-hp., 440-volt, 575-r.p.m. motor direct-connected to a 19.67 to 1 worm drive. This drives the main shaft, which in turn operates the 30-foot dumper through a 4.23 to 1 spur pinion and gear.

The two 60-inch primary inclined conveyors at Arthur are each 512 feet long and are driven by two 75 to 125 hp., 250-volt D. C. motors of 200 to 600 r.p.m., the speed being varied through rheostats. Each motor is equipped with a magnetic brake and direct connected through solid shaft to a pinion meshing with reduction gear on double pulley drive, located about 100 feet from the tail pulley. At Magna the two 54-inch inclined conveyors are each 397 feet long and are driven by a two-speed 60 or 125 hp., 440-volt, A. C., 295 or 575 r.p.m. motor connected through a flexible coupling to a double reduction enclosed herringbone gear. The first ratio is 6.9 to 1 and the second 5.8 to 1. The conveyor pulley is 83.25 inches in diameter. The feeders are driven from the same motors through sprocket gears from pulley shafts.

The two 54-inch level conveyors at Magna which handle the Symons products are 667 feet long and are driven by two 2-speed, 25 or 50 hp., 440-volt, 345 or 690 r.p.m. motors connected through Duckworth flexible couplings to double reduction enclosed gears. The first reduction is 4.5 to 1 and the second 6.6 to 1. The belt speed is either 175 or 350 feet per minute.

The 7-foot Symons crushers at both plants are driven by 250-hp., 440-volt, 450 r.p.m. motors direct connected through flexible couplings to the crusher pinion shafts. The gear reduction through crusher pinion and eccentric gear is 1.9 to 1.

The Symons crusher product plus the screen undersize at Arthur is delivered to two 36-inch inclined conveyors running east and west, thence to two north and south 36-inch inclined conveyors which deliver the ore to the center of the fine-ore bins. From here it is distributed over the lengths of the fine-ore bins by 32-inch horizontal tripper conveyors. Both sets of 36-inch conveyors are driven at a speed of 440 feet per minute.

At Magna the 54-inch level conveyors discharge to two 36-inch inclined conveyors, thence to two 42-inch inclined conveyors, which deliver the ore to the center of the fine-ore bins. From this point it is distributed over the lengths of the fine-ore bins by four 30-inch horizontal tripper conveyors.

Fine crushing

There are 12 and 13 sections of fine crushing at the Magna and Arthur plants respectively. Each Magna section consists of one 60-inch apron feeder, one 36-inch dry bucket elevator, nine 3 by 4 foot impact screens, and three sets of fine rolls, some 37-1/2 by 15 inches in size and the newer ones 44 by 16 inches. The feeder and rolls are driven by a 450-hp. 440-volt motor operated at 585 r.p.m. and connected through belt to a line shaft operated at a speed of 199 r.p.m. All rolls are driven by belts from the line shaft, a crossed belt being used for one shell. The impact screens and the dry 36-inch elevators are driven by a 75-hp. 440-volt motor, the elevator-head pulley being connected through a reduction gear unit and the screen cam shaft belted to the elevator-head pulley shaft extension. The Arthur sections consist of two 60-inch apron feeders, one 30-inch dry bucket elevator, six 3 by 4 foot impact screens, and two sets of rolls, some 37-1/2 by 15 inches in size and others 44 by 16 inch. At both plants the smaller rolls are being replaced by the 44 by 16 inch size as they break or wear out. At Arthur the feeders and rolls are driven by a 300-hp. 440-volt motor operated at 600 r.p.m., which is belted to a line shaft driven at 182 r.p.m., belt drives connecting the line shaft with the rolls and the feeders. The 30-inch dry elevators and impact screens at Arthur are driven by a 75-hp. 440-volt motor, operated at 875 r.p.m., connected by a flexible disk coupling to a double-reduction inclosed gear to head pulley running at 31 r.p.m. The elevator belts are 12-ply, are 105 feet long center to center, and carry 132 buckets 9 by 15 inches in size. Screen cam shafts are belted to extensions of the head pulley shafts. Both sizes of rolls are driven at a speed of 104 r.p.m. at Arthur, while at Magna the 37-1/2 by 15 inch rolls are driven at 110 r.p.m. and the 44 by 16 inch size at 98 r.p.m.

At both plants the rolls work in closed circuit with the impact screens, the circulating loads being 500 to 600 per cent of the new feed. Screen cloth ranges from 6 to 10 mesh in size, depending on the character and moisture content of the ore. The usual moisture content is 3 to 6 per cent, and the roll circuit is usually run dry but can be operated wet if the ore becomes too clayey and wet to screen properly. Sections at Magna with five ball mills operating are rated at 2,650 tons maximum and 2,250 tons average and the sections with four ball mills at 2,350 tons maximum or 1,950 tons average. The rated capacity per section at Arthur is 2,300 tons per day maximum or 1,900 tons average. More than rated maximum capacities have been handled at both plants, but at a sacrifice of fineness in grinding and metallurgical results.

Roll steel used in the form of shells for the fine rolls averages 0.075 pound per ton of ore milled, including the steel removed as worn-out shells.

Table 3 gives the screen sizes of the feed and final product of the fine-crushing sections.

Table 3.—Screen sizes of feed and final product of fine-crushing sections

Screen size	Mine run ore	No. 27 crusher product	Symons crusher product[1] Weight, per cent	Fine roll product[2] Weight, per cent
Plus 1.050 inch	Maximum size, 54 inches.	Maximum size, 6 inches.	3.98	-
Plus .742 inch			19.18	-
Plus .525 inch			15.81	-
Plus .371 inch			10.36	-
Plus 3 mesh........			8.67	-
Plus 4 mesh........			5.24	-
Plus 6 mesh........			4.34	-
Plus 8 mesh........			4.31	-
Plus 10 mesh......			3.60	0.52
Plus 14 mesh......			2.74	7.75
Plus 20 mesh......			2.11	12.43
Plus 28 mesh......			2.04	12.98
Plus 35 mesh......			2.21	9.96
Plus 48 mesh......			1.85	8.93
Plus 65 mesh......			2.19	6.76
Plus 100 mesh....			1.98	6.57
Plus 150 mesh....			1.46	4.00
Plus 200 mesh....			1.00	3.94
Minus 200 mesh..			6.93	26.16

1 - Feed to fine-roll circuit.
2 - Feed to ball mills.

Ball-mill Grinding

Both mills are divided into the same number of ball-mill sections as for roll crushing. Differences in the flow sheets in this department at the two plants and changes made at each in 1929 are shown by the diagram in Figure 5. The differences in the flow sheets of the ball-mill circuits at the two mills were occasioned by the difficulty of fitting new equipment into the spaces available and yet maintaining as simple a flow sheet as possible. The actual performance of the grinding and classifying equipment at the two mills has been almost identical.

At Magna each section consists of two 36-inch wet elevators, one or two (alternate) 8 by 14 foot primary drag classifiers, one or two (alternate) 7 by 10 foot primary ball mills, one 22-foot Dorr bowl-drag classifier, three secondary 7 by 10 foot ball mills, three 6 by 17-1/2 foot Utah Copper type drag classifiers, and three 8 by 17 1/2 foot Utah Copper type drag classifiers. At Arthur each section consists of one 36-inch wet elevator, two 15-foot Dorr bowl drag classifiers, one primary 7 by 10 foot ball mill, three 12 by 16 foot Utah Copper type drag classifiers and three 7 by 10 foot secondary ball mills.

At each plant the primary mills are loaded with balls 2 3/8 inches in initial diameter and the secondary mills with balls 2 inches in initial diameter. Balls are all manufactured at the Arthur plant foundry from scrap rail, scrap iron, and ferrochromium. They are cast in water-cooled molds and have an average Brinell hardness of 550. The ball loads in actual operation, with the mills filled almost to the lower edge of the open discharge trunnions, average about 42,000 pounds per mill. From 600 to 800 pounds of new balls are charged daily into each mill to maintain the loads at capacity. The mills are lined with 90-pound scrap rail cut in 4-inch sections and laid on end against the shell in concrete. The appearance of such a liner after having been in use a short time is shown in Figure 6. The inside diameter of the newly lined mills is 6 feet 4 inches and the effective length 9 feet 4 inches. The total consumption of balls per ton of ore milled is 1.27 pounds, and liner consumption amounts to 0.051 pound per ton milled.

The primary mills at Magna work in closed circuit with the 8 by 14 foot drag classifiers, and the secondary mills are in closed circuit with the 6-foot and 8-foot Utah Copper drag classifiers. Finished flotation feed pulp overflows both the 22-foot bowl classifier and the 6-foot and 8-foot secondary classifiers. The secondary classifier overflows go to a 225-foot Dorr thickener, and the 22-foot bowl overflow proceeds directly to flotation. At Arthur the primary ball mills are operated in open circuit, fed by sands from the 15-foot bowl classifiers, and the secondary mills are in closed circuit with the 12 by 16 foot Utah Copper drag classifiers. The 15-foot bowls and 12-foot drag classifiers overflow flotation feed, all feed going to Dorr tanks for thickening.

Although the impact-screen undersize is the feed to the ball-mill grinding department at both plants this material is classified before going to the mills. Actual daily sand loads to each mill average 1,000 to 2,000 dry tons, and the average moisture in the mill feeds is 25 to 30 per cent. Total classifier-overflow pulp ranges from 18 to 22 per cent of solids, which are increased to 30 to 35 per cent by the Dorr thickeners ahead of flotation treatment, the water thus removed being returned by pumps for reuse in classification. For purposes of estimating possible mill performance it is assumed that the fine rolls will produce an impact-screen undersize product averaging 35 per cent minus 100-mesh material and that from this feed each 7 by 10 foot ball mill will produce 270 to 290 tons of minus 100-mesh material per day.

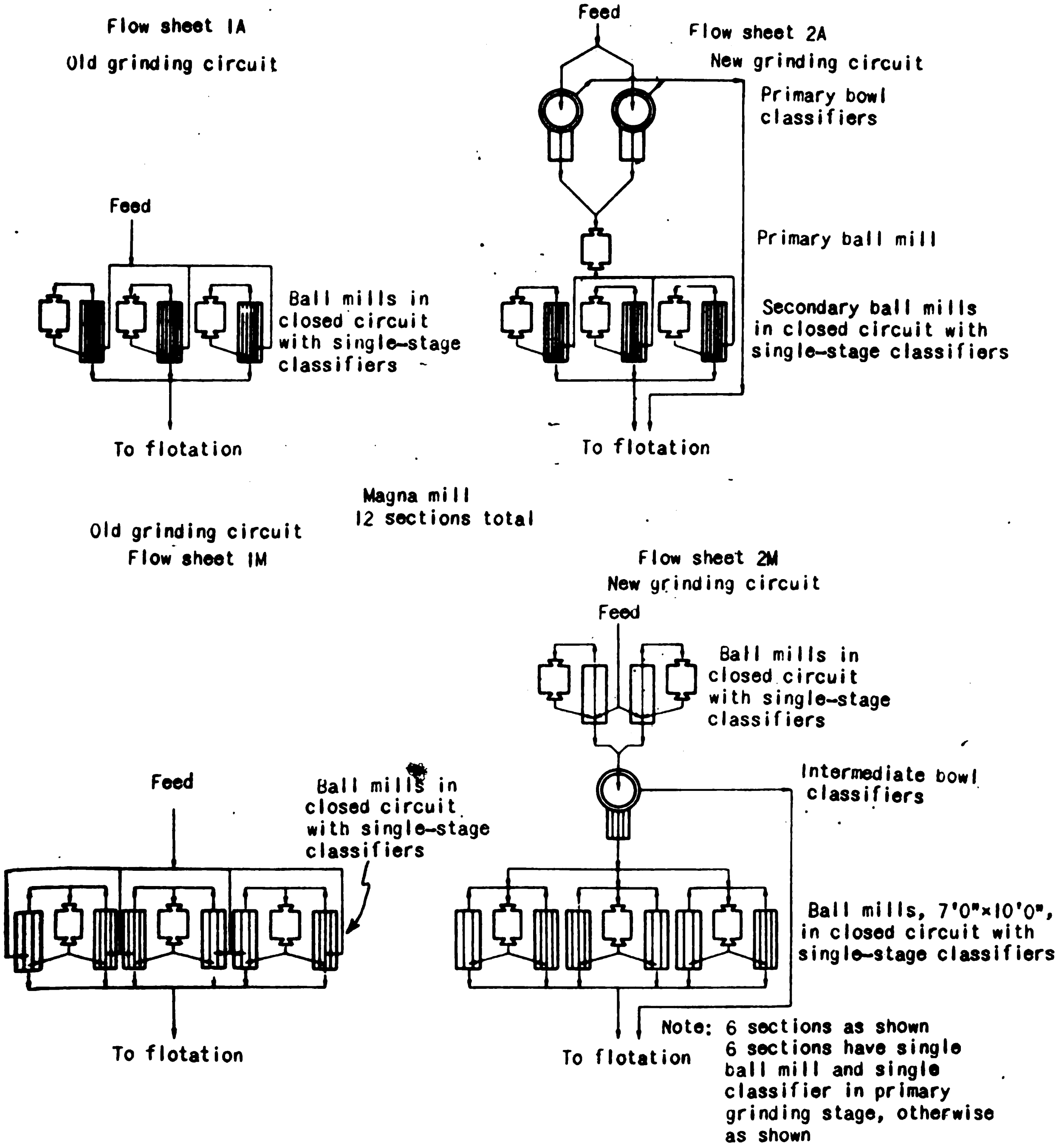

Flow sheet 1A
Old grinding circuit
Feed
Ball mills in closed circuit with single-stage classifiers
To flotation
Feed
Flow sheet 2A
New grinding circuit
Primary bowl classifiers
Primary ball mill
Secondary ball mills in closed circuit with single-stage classifiers
To flotation
Magna mill
12 sections total
Old grinding circuit
Flow sheet 1M
Feed
Ball mills in closed circuit with single-stage classifiers
To flotation
Flow sheet 2M
New grinding circuit
Feed
Ball mills in closed circuit with single-stage classifiers
Intermediate bowl classifiers
Ball mills, 7'0"×10'0", in closed circuit with single-stage classifiers
To flotation
Note: 6 sections as shown
6 sections have single ball mill and single classifier in primary grinding stage, otherwise as shown

Figure 5.- Flow sheets of grinding at the Arthur and Magma plants, from Dorr, J.V.N., and Marriott, A.D., Importance of Classification in Fine Grinding; Milling Methods, 1930: Trans. Am. Inst. Min. and Met. Eng., 1929, pp. 138 to 140

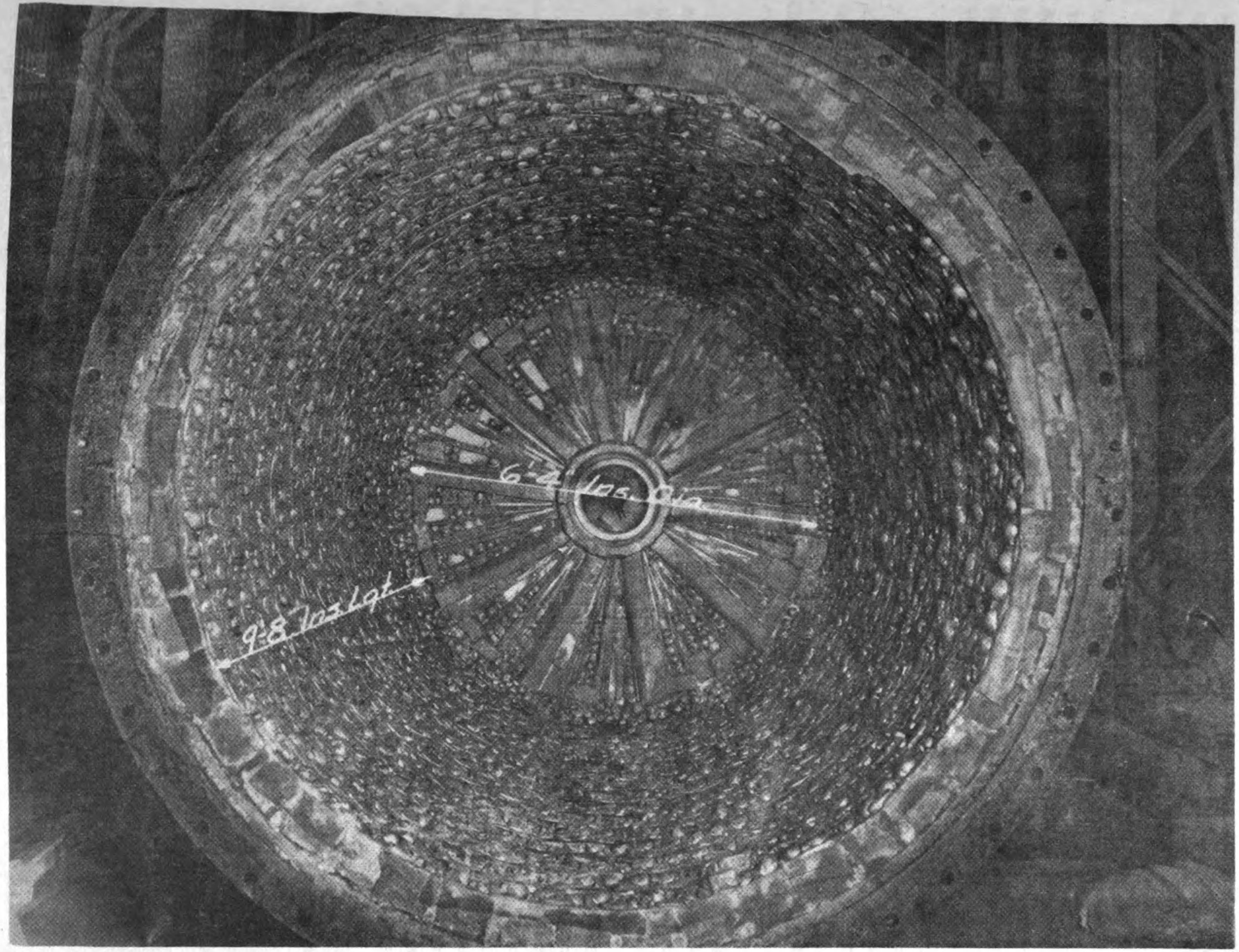


Figure 6.- Photograph of rail lined ball mill after 80 days in service

All the newly installed primary ball mills at both plants are driven by 200-hp. 440-volt synchronous motors operated at 164 r.p.m. and direct-connected to the ball-mill spur-pinion shaft. The ratio of pinion to ring gear is 8 to 1. These mills operate at a speed of 20.5 r.p.m.

The secondary ball mills at Magna are driven by 150-hp., 440-volt, 870-r.p.m., motors, connected with flexible couplings to double-reduction inclosed-gear units, which drive spur pinions at 158 r.p.m. The reduction ratio from pinion to ring gear is 8 to 1. Ball-mill speed under full load is 19.7 r.p.m. Each 36-inch wet elevator at Magna is driven by a 50-hp., 440-volt, 870-r.p.m. motor belted to a reduction-gear unit which drives the head pulley at a speed of 432 feet per minute. The belts are 12 ply, are 137 feet long center to center, and carry 178 buckets, each 9 by 18 inches in size.

The bowl rakes of the 22-foot bowl classifiers at Magna are driven at a speed of 2.8 r.p.m. by 5-hp., 440-volt, 720-r.p.m. motors through Texrope and worm-gear reductions, and the main rakes are driven at 16.9 strokes per minute by 10-hp., 440-volt, 600-r.p.m. motors with Texrope and spur-gear reductions. The six secondary Utah Copper drag classifiers of each section are driven at Magna by a 25-hp., 440-volt, 300-r.p.m. motor belted to a line shaft, which is belted to a twin-spur pinion. The rake speed of these classifiers is 16 strokes per minute.

Secondary ball mills at Arthur are driven by 150 and 200 hp., 440-volt, 900-r.p.m. motors connected by flexible couplings to double-reduction inclosed-gear units which drive the spur pinions at 158 r.p.m. The reduction ratio between pinion and ring gear is 8 to 1. The secondary ball mills operate at a speed of 19.7 r.p.m.

The 36-inch wet elevators at Arthur are each driven by a 75-hp., 440-volt, 720-r.p.m. motor, through double-reduction inclosed-gear units to the head pulley shaft. The belts are 120 feet long, are 12 ply, carry 150 buckets 9 by 18 inches in size, and travel at a speed of 432 feet per minute. The bowl rakes on the 15-foot Dorr bowl-drag primary classifiers at Arthur are driven at a speed of 3.5 r.p.m. by 3-hp., 900-r.p.m. motors through Texrope and worm-gear reductions. The main rakes are driven at 16.9 strokes per minute by a 10-hp., 600-r.p.m. motor through Texrope and spur-gear reduction. The Arthur secondary 12 by 16 foot drag classifiers are individually driven by 10-hp., 440-volt, 600-r.p.m. motors belted by Lenix drive to pulley on twin spur-gear shaft. The rakes of these classifiers operate at a speed of 20 strokes per minute.

Table 4 gives a screen-assay analysis of the fine-roll product which is fed to the primary ball mills after classification of the combined ball-mill products.

Table 4.—Screen assay analyses of ball-mill feed and final product, per cent

Screen size	Fine roll product or ball-mill feed		Ball-mill product or feed to flotation	
	Weight	Copper	Weight	Copper
Plus 10-mesh....	0.52	0.355	-	-
Plus 14-mesh....	7.75	.661	-	-
Plus 20-mesh....	12.43	.629	-	-
Plus 28-mesh....	12.98	.647	-	-
Plus 35-mesh....	9.96	.692	-	-
Plus 48-mesh....	8.93	.747	-	-
Plus 65-mesh....	6.76	.993	-	-
Plus 100-mesh..	6.57	1.358	9.60	0.197
Plus 150-mesh..	4.00	1.686	11.05	.271
Plus 200-mesh..	3.94	1.836	10.86	.684
Minus 200-mesh	26.16	1.289	68.49	1.257

Flotation

The flow sheets, Figures 3 and 4, give the major details of flotation treatment at each plant.

At Arthur 13 Dorr thickeners 75 feet in diameter, ahead of the flotation cells, dewater the pulp from 18 to 22 per cent of solids to 30 to 35 per cent of solids, the water being returned to mill mains by four 14-inch centrifugal pumps. The flotation cleaner tailings, without thickening, are mixed with the Dorr-tank underflow, thus lowering the actual percentage of solids in the feed pulp to the flotation machines to 27 to 30 per cent. All lime is added to the primary and the secondary ball-mill feeds, and all other reagents are added at the equalizer tanks marked No. 32 on Figure 3. Flotation machines used at Arthur are the double-spitz Janney mechanical-air machines, arranged in rows with one emulsifier followed by five frothing cells. In all, 103 rows of cells are used as roughers, the concentrates from all of these being collected in one sump and pumped to 11 rows of primary cleaners. On the primary cleaner units the emulsifier cells are also used as frothers, giving a total of 66 primary cleaner cells. Froth products from all the primary cleaner cells are collected in a sump and pumped to seven rows of secondary cleaners, five cells per row. Finished concentrates from the secondary cleaners are delivered by launder and elevator to Dorr thickeners. The primary cleaner tailings are returned by pump to join the original feed, and secondary cleaner tailings are returned by pump with the rougher concentrates to the head of the primary cleaners. Rougher tailings go to waste by gravity. The plant is normally rated at 25,000 tons per day but has handled as much as 34,000 tons in 24 hours with some loss in metallurgical results.

As before noted and as shown in Figure 4, the secondary classifier overflow at Magna is thickened in one 225-foot Dorr tank, and the 22-foot bowl classifier overflow goes directly to flotation treatment. The cleaner tailings at Magna are thickened in six 75-foot Dorr tanks before being returned to the head of the roughers. The combined classifier overflow ranges from 18 to 22 per cent of solids, and the thickening operation on secondary classifier overflow and cleaner tailings combined increases the solids in the feed pulp delivered to the roughers to between 24 and 26 per cent.

The main flotation plant at Magna consists of 60 rows of Janney Magna-type single spitz mechanical-air machines, each row consisting of 8 rougher cells, 2 primary cleaner cells, and 1 secondary cleaner cell. Some of the primary and secondary cleaners are not used at present. The same general flow is maintained as at Arthur, except that cleaning and recleaning operations are not handled on separate sections but by individual rows. Lime is added to the ball-mill circuits and part of each of the other reagents to the 225-foot Dorr-tank discharge at point 37 on Figure 4, the balance being added to the 22-foot bowl-classifier overflow at point 40 on Figure 4. About one-half of the total flotation feed comes from the 22-foot bowl classifiers and the other half from the 6 and 8 foot secondary drags. The 8-cell rows of rougher machines used at Magna have been found to handle more tons per cell than the shorter rows of 5 cells used at Arthur for equal metallurgical results. The Magna flotation plant handles efficiently 27,000 to 30,000 tons of solids per day, the consistency of the feed pulp amounting to 25 per cent of solids. The fact that more rougher cells in series make a more efficient flotation machine has also been demonstrated by an experimental row of 12 Janney mechanical-air cells at Magna. Some experimental work was also done at Magna on a 13-cell Janney straight mechanical machine, which was later transferred to Arthur for further testing of 16 rougher cells in series.

At both plants each flotation cell consists of an agitating compartment and a spitz, the bottom of the spitz being a sloping frame holding air mats. The motors are 7 1/2, 8, and 10 hp., 440-volt, 600-r.p.m. vertical type, direct connected to a shaft 31 inches long which carries two impellers, the upper being 20 inches and the lower 14-1/2 inches in diameter. The agitating cell is cylindrical in shape and is made either of silica brick or cast mastic, with four vertical baffles of adamant silica extending from the bottom to the lower edge of the upper impeller. Air is taken in either through stand-pipes or between the motor base and cell top. Incoming feed enters the cell through a tangential pipe at the bottom of the cell, and circulation of pulp from the spitz back to the agitating compartment is maintained through a pipe entering the center of the bottom of the cell. The air mats are 4-ply sewed Palma twill mounted on a cast-iron frame or pan, air-hose connections being made on the underside of the pan.

The Arthur plant has seven 10,000-cubic foot Roots rotary blowers, five ordinarily in use and two held in reserve as spares. Each blower is driven by a 275-hp., 440-volt, 600-r.p.m. synchronous motor connected to the blower by a Lenix belt drive. The Magna plant is equipped with five 10,000-cubic foot, one 20,000-cubic foot, and one 6,500-cubic foot Roots rotary blowers. As a rule two or three of the 10,000-foot blowers are idle as spares. The drives on all but the large blower are the same as at Arthur, the motor driving the 20,000-cubic foot blower being the 500-hp., 440-volt, 150-r.p.m. synchronous type. The air pressure at both plants ranges from 3 to 4.5 pounds per square inch at the mains. The pumps for the return of pulp at both plants are driven by direct connection through flexible couplings to 75-hp., 440-volt, 900-r.p.m. motors. The 14-inch pumps for return of water, shown as item 36, Figure 4, and item 31, Figure 3, are driven by 200-hp., 440-volt, 1,200-r.p.m. motors.

The discharge product from the 225-foot Dorr tank at Magna is returned to the flotation distributors by five 8-inch Wilfley sand pumps, each pump being driven by a 25-hp., 440-volt, 900-r.p.m. motor direct-connected through a flexible coupling to the pump impeller shaft. The Dorr tank discharge at Arthur goes by gravity to flotation. All Dorr tank sweeps on the 100-foot and 75-foot Dorr tanks at both plants are center driven by 5-hp. motors through back-gear and double worm reduction units or through Texrope and double worm reduction units. The 225-foot Dorr tank at Magna is of the traction type and is driven from the periphery of the tank.

The reagent feeders for milk of lime, sodium aerofloat solution, cyanide solution, and raw cresylic acid are rectangular steel tanks containing vertical steel disks mounted on a horizontal shaft. Cast-iron cups are bolted to the outer edges of the disks. The feeders are usually driven by 1/4 or 1/2 hp., D. C. motors, connected to the feeder shaft through a worm reduction gear. The rate of reagent addition is varied by a rheostat located at the head of the flotation machines or by changing the number of cups on the disks.

A lime-burning plant is located at the Magna mill. Lime rock, crushed to 1-inch size and with the minus 1/4-inch material removed by screens, is shipped to Magna from the Topliffe quarry of the American Smelting & Refining Co., 20 miles south of Salt Lake. Two rotary kilns, each 125 feet long, one 6 feet and the other 8 feet in diameter, are so arranged that they may be used as one kiln 250 feet long or as individual kilns if greater capacity is needed. The kilns are fired with powdered coal, and a ratio of 1 coal to 3.7 to 4.0 burned lime is maintained. The burned lime averages 77 to 80 per cent of available or water-soluble CaO and is slaked at the lime plant. Two Hardinge ball mills are fed with the dry-burned lime and water, the ground product going to two drag classifiers which return the coarser particles to the mills. The classifier overflow, containing about 20 per cent of CaO in solution and suspension, goes to sumps and from there is pumped to storage tanks at Arthur and Magna through 4-inch lines. The storage tanks are equipped with mechanical agitators to prevent settling of the solids, and the milk of lime is pumped to feeders located above the storage tanks. An excess of milk of lime is continually pumped to the feeders, the overflow returning by gravity to the tanks. The cost of burned lime delivered to the tanks as milk of lime ranges from $6 to $8 per ton, depending mostly on the quantities used by the mills.

All the sodium-ethyl aerofloat used at the two plants is manufactured at Arthur. Absolute denatured ethyl alcohol is treated with phosphorus pentasulphide in a chrome-nickel steel still or kettle, and when the reaction is complete the resultant thio acid is drawn into a solution of caustic soda in water, a slight excess of caustic being used. The final solution is slightly alkaline and carries 2.638 pounds of reagent per gallon. In this form it is piped by gravity to the Arthur storage tanks and shipped to storage tanks at Magna in a tank car, from whence it is piped to the feeders.

The average flotation reagent consumptions are as follows:

	Pounds per ton of ore milled
Burned lime	[1]3.79
Sodium aerofloat	.017
Raw cresylic acid	.18
Aero brand cyanide	.05

1 - Sufficient to produce pH value of 8.8 to 9.2 in flotation feed water.

An experimental flotation plant is maintained at Arthur, where many types of flotation machines and hundreds of reagent combinations have been tested out on full mill scale. The experimental plant at present contains one 400-ton capacity Janney straight mechanical machine, consisting of one emulsifier cell, eight rougher cells, one primary cleaning, and one secondary cleaning cell; one 900-ton capacity Janney straight mechanical machine consisting of one emulsifier, sixteen rougher, two primary cleaner, and one secondary cleaner cells;

one 450-ton capacity Janney Magna-type mechanical-air machine consisting of one emulsifier, eight rougher, one primary cleaner, and one secondary cleaner cells; and one Southwest air-lift type machine rated at 400 tons capacity per day.

Some large-scale test work has been done in the Arthur experimental plant on the separate treatments of the primary bowl-classifier overflow product, which contains most of the primary mine slime, and the secondary drag-classifier overflow product. The results recorded so far are no better than can be obtained by treating the mixed primary and secondary classifier-overflow products.

Tests so far made with other types of flotation machines, which use less power than the Janney type on Utah Copper ore, have not shown any over-all saving when considered from the standpoints of metallurgical results and consumption of reagents.

The metallurgical results obtained at the Arthur and Magna plants are practically the same and depend upon the tonnage treated, which in turn governs the fineness of grinding.

DEWATERING OF CONCENTRATES

At the Arthur plant there are two 100-foot and four 75-foot Dorr thickeners for concentrates and five 4-leaf 13-foot American filters. The concentrates, averaging about 25 per cent of solids, are elevated to feed the Dorr tanks by three 36-inch bucket elevators; and the thickened concentrates, averaging 55 to 65 per cent of solids, are again elevated by two 18-inch bucket elevators and distributed by launders to the filter basins. The average filter cake carries 9 to 10 per cent of moisture. Usually only one 100-foot thickener is actually in use, the overflow from this passing through the 75-foot tanks, thence through the old concentrates bins to a settling pond. All overflow water and filtrate water are sent to waste through the settling pond. This pond and the bins are drained once a year, and the accumulated sediment is returned by pump to one of the Dorr thickeners. Most of the material collected in the settling pond comes from broken filter bags.

The flow sheet for dewatering of concentrates at Magna is very much the same as at Arthur. Two 75-foot thickeners and six 13-foot 4-leaf American filters are in use. Solids in feed to tanks, tank discharge, and moisture in filter cake are the same as at Arthur. All tank overflow and filter water are passed through bins and the settling pond to waste.

Vacuum at Arthur is furnished by three 27 by 14 inch vacuum pumps and one 36 by 20 inch pump. The Magna plant has two small-size and two large-size vacuum pumps. Actually, under present conditions, only one small pump or at times one large pump is in use at each plant. The small pumps are driven by 112-hp., 440-volt, 900-r.p.m. and the large pumps by 275-hp., 440-volt, 600-r.p.m. synchronous motors, all pumps being driven by belts through Lenix drives. In these departments all conveyors shown in Figures 3 and 4 are driven by direct-connected motors, through inclosed-gear reducing units with drive pulleys at different points.

DISPOSAL OF TAILINGS

Tailings from both mills run north by gravity into an artificial pond about 10 square miles in area. A rock dam or dike extends for 10 miles around the west, north, and east sides of the pond, and a standard-gage railroad track is maintained all the way around for additions and repairs. Most of the fill has been made of rock waste from the Utah Copper mine at Bingham, although adjacent dirt has been used in some places. Most of the final overflow water goes out of the pond through weirs into a canal flowing in a northwesterly direction into Great Salt Lake. The upkeep and raising of the dike require a large and constant expenditure.

With tonnages from the mills increasing constantly it was foreseen in 1916 that the tailings pond then in use would have to be enlarged. Accordingly it was arranged with the Los Angeles and Salt Lake and Western Pacific Railroads to move their main tracks 1 to 1-1/2 miles northward from their former location along the northern edge of the old tailings dike to an east-west line 9 miles long. This move, necessitating building many miles of new dike as well as moving of the railroad tracks, was completed in 1918, and more than doubled the area for tailings disposal.

WATER SUPPLY

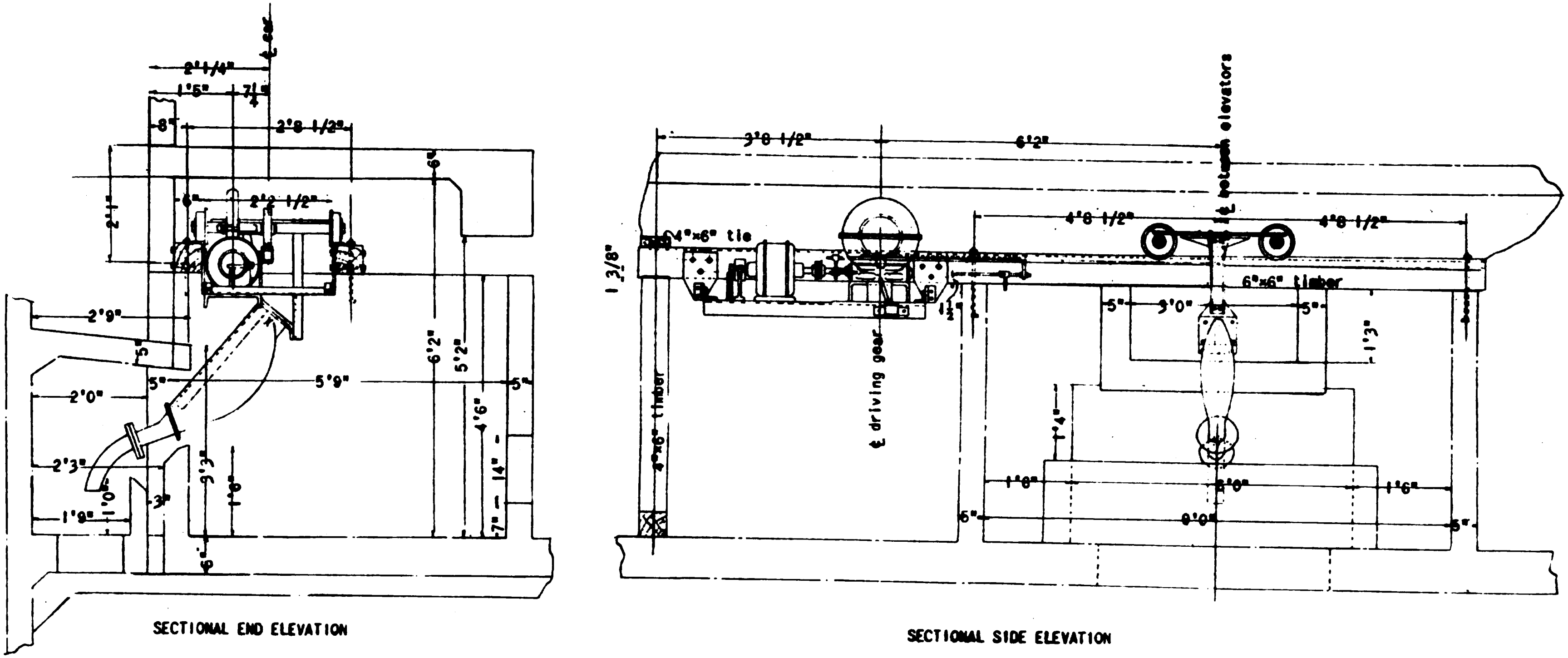
As previously noted the main mill-water supply is obtained from Utah Lake through a canal and pumps to the reservoirs. This water contains fairly large amounts of solid matter in solution and some material in suspension, and is never entirely clear. The solid matter in solution is principally bicarbonates, sodium chloride, and calcium and magnesium salts. It is estimated that 0.19 pound of burned lime is consumed by each ton of new water in reacting with the soluble salts in the water before the alkalinity of the water is increased to the point now carried in mill practice. Hard, rocky deposits of calcium carbonate formed by interaction of water and lime, are found on elevator buckets and launders and at other points in the mill. The water from the Utah canal is slightly alkaline, having a pH value of 8.3 to 8.4. The water shows some seasonal variation in composition, the soluble solid content being considerably higher in hot, dry weather and lower in winter or spring.

An auxiliary water supply has been developed from lakes about 6 miles east of the mills. This water is partly from springs but principally from the drainage of a large area to the lakes. The Riter canal connects the lakes to a pumping station at the old powerhouse at Magna. When ice chokes the Utah Lake canal or the Utah canal is shut down for cleaning the Riter canal water is pumped to the mill reservoirs, the pump lift for Riter water being 212 feet greater than that for Utah Lake water. The Riter canal water contains a greater percentage of solid matter in solution than the Utah water does.

No water is reclaimed for reuse except that which overflows the Dorr thickeners ahead of flotation, although it is possible to return a considerable quantity of tailings water from the tailings pond to pumps through the Riter canal.

WEIGHING AND SAMPLING

The products of the Symons cone crushers are automatically weighed on the conveyor belts by Merrick weightometers, at point 16, Figure 4, and point 14, Figure 3. Moisture samples are taken immediately after the products pass over the weightometers, and from these the dry tonnages are calculated. At Arthur the heads sample is cut from the 1-inch material entering the fine-ore bins and at Magna from the wet pulp after final grinding. Samples of tailings and concentrates are cut from the pulp streams at points shown in Figures 3 and 4. All samplers, including those used for experimental work, are electrically driven and electrically timed. All sample cutters are controlled from one master timer at each



SECTIONAL END ELEVATION

SECTIONAL SIDE ELEVATION

Figure 7. – General arrangement of sample cutters

plant, which operates the cutters every 10 minutes, except the Arthur main-heads sampler, which is operated every 5 minutes from a separate timer. The width of the opening in the cutter varies for different samples but is not less than 5 times the diameter of the largest ore particle to be sampled. The general arrangement of the cutters is shown in Figure 7. At points where a pulp sample is cut and the stream is large, the cutter discharges into a revolving bowl from which four pipes discharge the sample over fixed openings below. The size of the sample is thus reduced to any desired amount before it is collected in buckets. The buckets are carried to the sample room once or twice every 8-hour shift. All pulp samples are weighed wet, then dried on steam hot plates and the dry weights taken, so that the content of moisture or solids may be calculated. The Arthur mill heads sample, 1 inch in maximum size, is collected for 24 hours in a steel bin, from which it is run every morning over a Vezin sampler which makes a 10 per cent cut, thence through small crushing rolls, and through a second Vezin cutter which also makes a 10 per cent cut. The final sample, 1/4 inch in size and 600 to 1,000 pounds in weight, is riffled by hand to about 10 pounds. This 10-pound sample is dried, passed through rolls, and riffled down to between 300 and 500 grams the final sample being bucked through a 100-mesh screen and sent to the assay office.

Other samples are riffled to convenient size, after drying, then bucked through a 100-mesh screen and sent to the assay office. Certain riffled cuts of each sample, representing tonnage cuts of each, are collected in cans for the entire month, then rolled and a sample riffled out and bucked for assay to check the weighted daily averages. These samples are called monthly composites, and the heads and tailings usually check the sheet average to less than 0.01 per cent of copper. Composites of concentrates usually check to less than 0.15 per cent of copper.

Mill sampling, weighing, and analytical work are checked further by the smelter returns. The concentrates are shipped to the Garfield smelter of the American Smelting & Refining Co., where they are sampled for moisture, weighed on railroad scales, and sampled for assay by an automatic device as unloaded. Mill and smelter assays on duplicate control samples from each lot of 8 cars must check within 0.12 per cent of copper, or an umpire assay is made on a third sample. The total difference between smelter-settlement returns and mill indicated recoveries over a period of 8 years is less than 0.005 per cent of copper.

All analytical work is done at the assay office at Arthur, prepared samples being sent by truck from the Magna plant sampling department. All mill samples, as well as control pulps, are analyzed for copper by the iodide method. Triplicate determinations are made on general samples of heads, tailings, and concentrates, including both shift and 24-hour composite samples.

CONVEYORS AND ELEVATORS

Data on the larger conveyors and elevators have been given in the descriptions of equipment for the various departments; but all details as to size, slope, and elevation of ore by each belt are obviously beyond the scope of this paper. Although gravity flows were planned wherever possible in the original designs of the mills the rapid changes made in the flow sheets in succeeding years have necessitated the use of more mechanical elevation of products. Such practice is cheaper than attempting to rearrange all other apparatus to maintain a gravity flow. For instance, the actual cost of elevating ore 60 feet vertically by the 36-inch wet-bucket elevators at Arthur, shown by item 25 on Figure 3, is as follows:

	Cost per ton of ore milled
Labor......	$0.00061
Supplies	.00183
Shops......	.00047
Power......	.00262
Total......	$ .00553

The 36-inch elevators actually handle 5 tons of ore and water for every ton milled, due to the circulating load and water content, so that the cost per ton of material handled is only $0.001106.

CONCENTRATOR RECOVERY

Average concentrator results are shown in Tables 5 and 6, which give assays of heads and products and general metallurgical data, respectively, for April, 1930.

Although the figures in Tables 5 and 6 are about average at present, cleaner tailings and higher recoveries of copper, gold, and silver could be made with lighter tonnages and finer grinding. In June, 1930, the Magna plant made tailings containing 0.0674 per cent of total copper, the lowest copper content ever recorded at these plants. The recovery of total copper for June, 1930, was 93.23 per cent, and the recovery of sulphide copper amounted to 95.11 per cent. The average ore treated in June was 25,987 tons per day and the tailings product contained 7.88 per cent plus 100-mesh material.

Tables 7a and 7b show the relation of copper recovery and grade of concentrates to fineness of grinding. The results in these tables were obtained from 30-day composite samples taken at the Magna plant in 1929.

The comparisons in Table 7a and 7b show that it costs $0.05982 more per ton to grind flotation feed to 9.60 per cent plus 100-mesh size than to grind to 26.13 per cent plus 100-mesh size. The increase noted is not all due to grinding costs, as overhead and other items remain fixed for any tonnage treated. However, with finer grinding copper recovery increases from 82.13 to 90.67 per cent, and the grade of concentrates increases from 30.26 to 31.94 per cent.

Figure 8 gives in graphic form the results obtained in the laboratory by varying the degree of grinding on three different samples of average Utah Copper ore. In these tests only copper recovery was sought. This figure shows that it is possible to overgrind and thereby ruin flotation results, but such grinding is not likely to be attained in the plants. These results and other laboratory work indicate that the ultimate economical degree of grinding at the Utah Copper mills is all minus 100-mesh size and about 85 per cent minus 200-mesh size.

Detailed microscopic analyses of sulphide minerals in various screen sizes of samples of tailings and concentrates are shown in Table 8. The same samples were used for the microscopic analyses of Table 8 and for the sizing and chemical analyses given in Table 7a.

The minus 200-mesh size material for the microscopic work was elutriated into four successive sizes, as shown in Table 8, and although some increase in copper recovery is shown below the minus 25-micron size the most marked change is from the plus 200-mesh size to the plus 25-micron size. This indicates that grinding of minerals should be carried to the minus 200-mesh and plus 25-micron sizes.

Table 5.—Tonnages and average assays of concentrator heads and products of the Magna plant for April, 1930

	Tons	Assays, per cent						Assays, ounces per ton	
		Copper		Iron		Insoluble	Sulphur	Gold	Silver
		Total	Nonsulphide[1]	Total	Nonsulphide				
Heads	843,800	0.9572	0.035	2.65	0.25	-	-	0.0090	0.0884
Concentrates	22,926	31.4880	-	26.01	-	5.29	35.20	.242	2.56
Tailings	820,874	.1046	.030	2.27	.25	-	-	.0025	.0195

1 - The nonsulphide copper content is determined by rolling 2 1/2-gram samples for two hours with 100 ml. of a 4 per cent water solution of sulphur dioxide, filtering and determining the copper in the filtrate by the iodide method.

Table 6.—Metallurgical data at the Magna concentrator for April, 1930[1]

Total ore milled, tons	843,800
Total ore dumped, cars	10,458
Average ore per car, dry tons	80.62
Moisture in mill feed, per cent	5.078
Average ore milled per day, tons	28,127
Operation per day, hours	24
Average number of crushing and grinding sections in actual operation	11.75
Average ore treated per section per day, tons	2,394
Plus 100-mesh material in tailings, per cent	11.76
Total concentrates produced, tons	22,926
Average concentrates produced per day, tons	764
Recoveries, per cent	
Total copper	89.37
Sulphide copper	92.14
Nonsulphide copper	16.61
Gold	72.98
Silver	78.54
Ratio of concentration, tons into 1	36.805
New water per minute per ton of ore milled per day, gallons	0.614
Solids in feed to filters, per cent	56.77
Average number of American filters in operation	5.07
Average concentrates handled per day per filter, dry tons	151.22
Average moisture content of filter cake, per cent	9.924
Average filtering area per ton per day, square feet	4.583

1 - The Arthur plant shut down during this period.

Table 7a.--Screen-assay analyses showing recovery and grade of concentrates following coarse grinding[1]

(Total cost of direct milling with this fineness of grinding = $0.24235 per ton;

rate of milling = 27,643 tons per day)

Screen sizes, mesh	Heads, calculated			Actual tailings			Actual concentrates			Recovery, per cent
	Material, grams	Copper		Material, grams	Copper		Material, grams	Copper		
		per cent	Grams		per cent	Grams		per cent	Grams	
Plus 100..	2,613.17	0.3894	10.17584	2,599.66	0.2750	7.14906	13.51	22.404	3.02678	29.74
Plus 150..	1,016.09	.9797	9.95440	984.97	.2000	1.96994	31.12	25.657	7.98446	80.21
Plus 200..	945.37	1.1216	10.60237	911.00	.1250	1.13875	34.37	27.536	9.46412	89.26
Minus 200	5,425.37	1.2475	67.67898	5,237.28	.1400	7.33219	188.09	32.084	60.34679	89.17
Composite	10,000.00	.9841	98.40880	9,732.91	.1807	17.58737	267.09	30.260	80.82143	82.13

Table 7b.--Screen-assay analyses showing recovery and grade of concentrate following fine grinding[1]

(Total cost of direct milling with this fineness of grinding = $0.30217 per ton;

rate of milling = 24,017 tons per day)

Screen sizes, mesh	Heads, calculated			Actual tailings			Actual concentrates			Recovery, per cent
	Material, grams	Copper		Material, grams	Copper		Material, grams	Copper		
		Per cent	Grams		Per cent	Grams		Per cent	Grams	
Plus 100..	960.09	0.1969	1.89026	956.52	0.1410	1.34869	3.57	15.170	0.54157	28.65
Plus 150..	1,104.94	.2707	2.99131	1,096.50	.0950	1.04167	8.44	23.100	1.94964	65.18
Plus 200..	1,085.74	.6841	7.42794	1,061.50	.0800	0.84920	24.24	27.140	6.57874	88.57
Minus 200	6,849.23	1.2571	86.10016	6,606.19	.0900	5.94557	243.04	32.980	80.15459	93.67
Composite	10,000.00	.9841	98.41085	9,720.71	.0945	9.18607	279.29	31.947	89.22478	90.67

[1] - Martin, H. S., Microscopic Studies of Mill Products as an Aid to Operation at the Utah Copper Mills: Am. Inst. Min. and Met. Eng., Tech. Pub. 255, p. 20.

Table 8.—Microscopic determination of distribution of sulphide minerals in concentrates and tailings in composite 30-day samples[1]

Minerals	Per cent						
	Plus 100-mesh	Minus 100 plus 150 mesh	Minus 150 plus 200 mesh	Minus 75 plus 25 microns	Minus 25 plus 10 microns	Minus 15 plus 5 microns	Minus 7 plus 3 microns
Concentrates							
Pyrite	29.85	29.26	23.22	12.35	4.40	1.40	0.75
Chalcopyrite	55.60	53.77	59.97	66.31	71.17	82.73	87.54
Bornite	2.68	4.36	6.97	5.64	4.18	2.40	0.98
Chalcocite	7.65	8.95	5.91	8.62	9.24	6.59	4.07
Covellite	4.22	3.65	3.93	7.07	11.24	6.86	6.66
Total	100.00	99.99	100.00	99.99	100.23	99.98	100.00
Tailings							
Pyrite	88.00	89.10	90.30	96.50	97.3	97.45	96.50
Chalcopyrite	7.47	8.13	6.42	2.45	1.9	1.65	2.07
Bornite	1.91	0.48	0.65	0.11	0.1	0.10	0.10
Chalcocite	2.50	2.02	2.48	0.51	0.45	0.34	0.68
Covellite	0.55	0.24	0.22	0.37	0.37	0.42	0.60
Total	100.43	99.97	100.00	99.93	100.12	99.96	99.95

1 - Martin, H. S., Microscopic Studies of Mill Products as an Aid to Operation at the Utah Copper Mills: Am. Inst. Min. and Met. Eng., Tech. Pub. 255, p. 10.

COSTS

Table 9 summarizes concentrator costs, Table 10 gives the costs segregated into labor, supplies and power, and Table 11 shows the distribution of power by departments. The costs in these tables are monthly figures for 1929, when both the Arthur and the Magna plants were in operation and give a more normal cost distribution than could be evolved for 1930, when the Magna plant absorbed the entire overhead costs.

Table 9.—Summary of direct milling costs for 29 days of operation in 1929
(Ore treated, 1,406,300 tons; concentrates produced, 37,344 tons)

	Operating labor	Mill repair labor	Power	Supplies	Reagents	Shops labor and supplies	Miscellaneous[1]	Total
Coarse crushing, mine run to 1 inch	$0.00570	$0.00451	$0.00407	$0.00305	–	$0.00224	$0.00013	$0.01970
Fine crushing, 1 inch to 8 mesh	.00551	.00791	.01917	.01002	–	.00225	.00013	.04499
Grinding, 8 mesh to 10 per cent plus 100 mesh	.00518	.00491	.04974	.02920	–	.00276	.00004	.09183
Flotation	.00584	.00502	.03262	.00956	$0.03527	.00287	.00017	.09135
Dewatering of concentrates	.00159	.00098	.00118	.00089	–	.00041	.00019	.00524
Assaying and sampling	.00215	–	.00017	.00163	–	.00010	.00012	.00417
Experimental and research	.00237	–	.00118	.00339	–	.00005	.00021	.00720
Disposal of tailings	.00324	–	–	.01073	–	–	.00025	.01422
Initial water supply	.00609	–	.00225	.00117	–	–	.00048	.00999
Administration and miscellaneous	.00524	–	–	.00280	–	.00076	.00468	.01348
Total	$0.04291	$0.02333	$0.11038	$0.07244	$0.03527	$0.01144	$0.00640	$0.30217

1 - Includes some labor through special accounts.

Table 10.—Distribution of costs into total labor, supplies, and power

	Cost per ton milled			
	Total labor	Supplies	Power[1]	Total
Plant administration and general[2]	$0.02530	$0.01795	$0.00581	$0.04906
Coarse crushing	.01227	.00319	.00424	.01970
Fine crushing	.01548	.01017	.01934	.04499
Ball mill grinding	.01261	.02937	.04985	.09183
Flotation concentration	.01350	.04502	.03283	.09135
Concentrate dewatering	.00307	.00093	.00124	.00524
Total	$0.08223	$0.10663	$0.11331	$0.30217

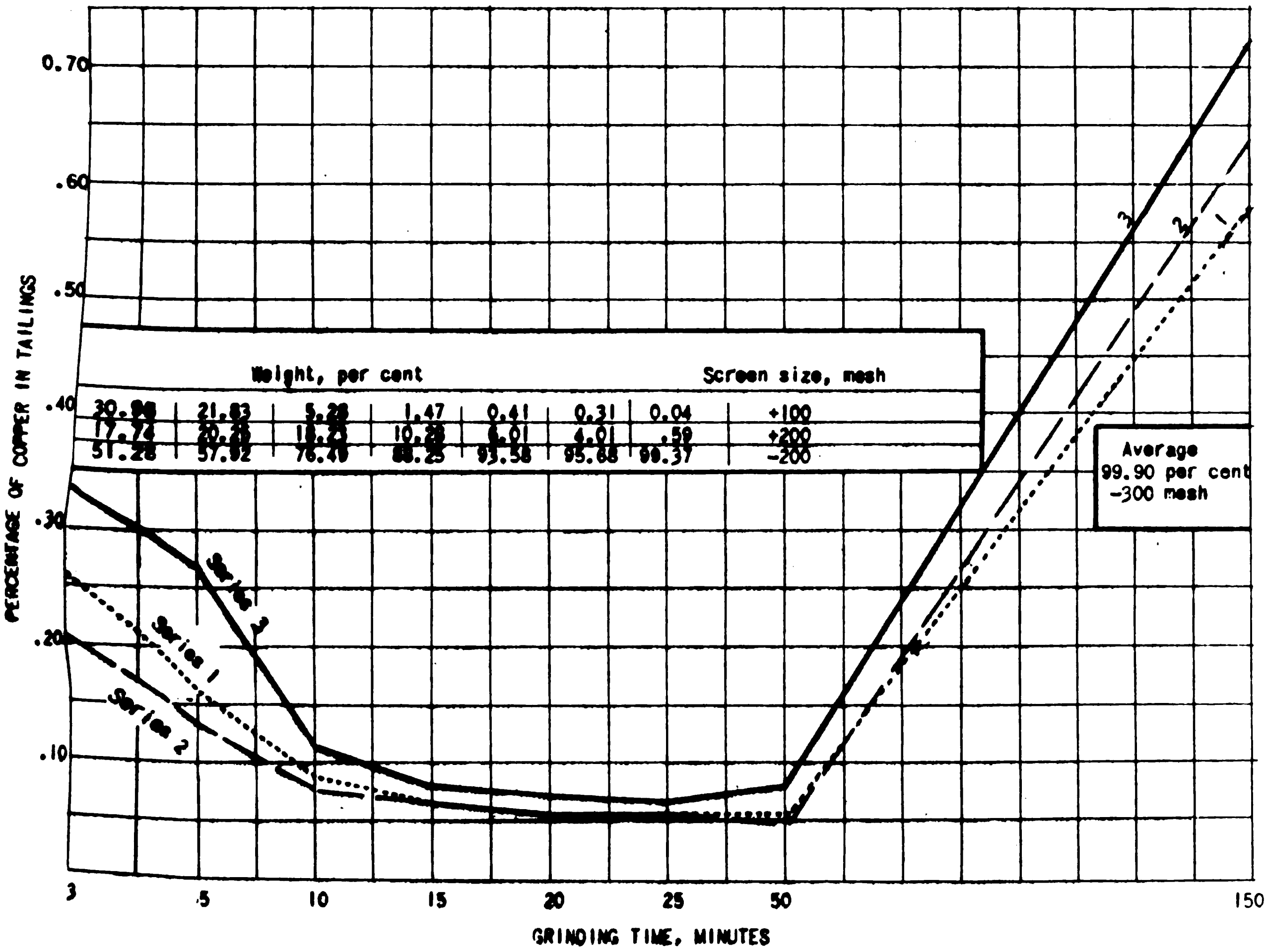
1 - Including power used by shops and other auxiliary accounts, cost of which is distributed to various departments.

2 - Including assaying, sampling, research, experimental work, tailings disposal and initial water supply.

Table 11.—Distribution of power

	kw. h. per ton milled	Per cent of total power
Coarse crushing	0.61	3.67
Fine crushing	2.89	17.38
Ball-mill grinding	7.51	45.16
Flotation	4.92	29.59
Dewatering of concentrates	.18	1.08
Experimental and research	.18	1.08
Initial water supply	.34	2.04
Total direct milling	16.63	100.00
Shops, foundry, compressors, houses, assay office, administration, etc	.47[1]	
Grand total	17.10	

[1] - Shown in costs (Table 9) under assaying and sampling, administration, and miscellaneous. Cost actually distributed as miscellaneous to various departments.



Weight, per cent							Screen size, mesh
30.98	21.83	5.28	1.47	0.41	0.31	0.04	+100
17.74	20.25	18.23	10.28	6.01	4.01	.59	+200
51.28	57.92	76.49	88.25	93.58	95.68	99.37	-200

Figure 8.- Effect of grinding on flotation recovery